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102-06540/

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MRL 95-041 (OPJ) June 1995

For Presentation at
The 6th High-Tech Seminar:
State-of-the-Art Blasting Technology, Instrumentation and Explosives Applications,
Organized by **Blasting Analysis International, Inc.**
Boston, Massachusetts, USA, July 8-14, 1995.

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The Mechanisms of Rock Damage in Blasthole Open Stope Mining: Blast Induced versus Stress Induced

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Abstract:

A large scale blast damage project was carried out over the past two years at INCO Ltd Manitoba Division co-funded by the Canada/Manitoba Mineral Development Agreement. The experiment was conducted at the T-3 Mine in two mining stopes with an experimental drift developed in the hanging-wall for the installation of monitoring instrumentation. The objective was to observe and subsequently to reduce ore dilution by monitoring the performance of blasts, the hanging wall movement, as well as the cavity of the open stopes.

The techniques and instrumentation used in this research project include geomechanical assessment of the rock mass, Optech Cavity Monitoring System (CMS), vibration monitoring using B&K accelerometers and OYO high frequency geophones, VOD measurement using VODEX-100 system and Slif-X VOD system, borehole camera and extensometers, cable strain measurement and vibrating wire load cells, numerical modelling of stress redistribution and damage predictions, Boretrak borehole deviation measurement, Probex dilatometer for in-situ rock mass modulus measurement and fragmentation analysis.

The excavation of the two experimental stopes, completed in 21 blasts, was fully monitored by the comprehensive instrumentation. Blast damages as identified by the borehole camera surveys, extensometer readings and CMS scans were correlated with blast designs and near-field vibration levels. The types of ground damage were separated between ground stress induced and blast induced by means of hanging-wall observations and a 3-D numerical modelling program. The paper presents the methodology employed in the instrumentation and summarizes some of the results for analyzing rock damages induced by blast and by stress. Guidelines to help minimize the blast induced damage are given considering the principles of blast design and blast damage mechanisms. The importance of studying rock geomechanics in correlation with blast damage is strongly emphasized.

Figures follow the text.

1. INTRODUCTION

The project entitled "A Study of Blasthole Stope - Blast Damage Control" (INCO, 1994) is one of the most comprehensive studies in this field which was carried out directly by the mining industry. The project was co-funded under the Canada/Manitoba Mineral Development Agreement with INCO Ltd Manitoba Division. The experiment was conducted at T-3 Mine by INCO Mines Research Department from 1992 to 1994. The consensus between INCO Mines Research and the government scientific authority was that research was needed to be performed in this area of science in order to provide underground blast designers with information to help minimize blast damage. Blast damage must be controlled to help reduce dilution of waste, ore and backfill from the peripheral walls of blasthole stopes for economic and safety reasons.

T-3 Mine was chosen for conducting the comprehensive stoping performance analysis of Block 19 and 21 on the 2850 Level. For this project, a hangingwall drift was driven to provide access for monitoring the two primary blocks. Geomechanical assessment of the rock mass was completed prior to the experiment. The following instrumentation and techniques have been utilized in this research project:

- Optech Cavity Monitoring System (CMS);
- Vibration Monitoring using B&K accelerometers and OYO geophones;
- VOD measurement using both Vodex-100 and Slif-X systems;
- Borehole camera and extensometers;
- Cable strain measuring devices and Vibrating wire load cells;
- Boretrak borehole deviation measurement;
- Probex dilatometer;
- Fragmentation analysis.

The total dollar value of this project was $313,000. The people involved included the Operations Group from the mine in conjunction with the Blasting/Geomechanics Group of the Mines Research Department of INCO Manitoba Division.

The aim of this project was to gain knowledge that mining engineers can use to design blasts that control damage to the peripheral walls of blasthole stopes. The results that this research project planned to generate were to determine what effect blast design parameters had on the creation of blast induced damage and to determine if blast vibration measurement could be used to estimate blast induced damage in blasthole stopes.

The hangingwall of the stopes is often the main rock wall of concern. Therefore, a drift was developed into the hangingwall to provide a workplace for monitoring purposes. The blast damage observed in the hangingwall with the use of a borehole camera, extensometers and dilatometer was the creation of new fractures, the opening of existing fractures, the displacements of the rockmass and the change in the modulus of the rock. By analyzing these observation results over the mining period (about 3 months) before and after each blast, the

effect of blast induced and stress induced damages could be distinguished while considering the size of blasts and stress redistributions.

The mining of two primary stope blocks within the same stope complex and in close vicinity to one another was monitored. In this way, the stress redistribution around the stopes could be modelled with a similar ground stress field and the geological properties of the rockmass would be similar. This would help in reducing the factors affecting damage other than blasting. The blasts were all designed using the slot/slash mining method where a vertical slot is established in the ore providing an opening to blast into. The holes were drilled from a drill drift excavated above the ore to be mined. All holes were drilled in a fanned pattern in a downward direction into the orebody.

Digital blast monitoring instruments were used to measure blast vibrations and explosives performance. Blast vibrations were monitored in the immediate vicinity of blasting using accelerometers and high frequency geophones. Computer software was used to examine the acceleration, particle velocity and frequency content of the recorded waveforms.

In order to separate blast induced damage from ground stress induced damage, ground stresses were measured and modelled. A three dimensional boundary element stress model was used to analyze stress redistribution. Load cells installed on the ground support bolts were monitored to measure changes in ground stress. In addition, blasted ore was photographed during mucking for analyzing the size distribution of fragmentation.

2. EXPERIMENT LAYOUT

2.1. Geomechanics of the Rockmass

The Thompson Nickel Belt consists of a band of metamorphosed Proterozoic sediments underlain by Archean basement gneisses. The Thompson ore zone is located within the Proterozoic package and consists of massive to disseminated sulphide with varying amounts of inclusions. The inclusions are typically ultramafic and/or lithic fragments of the wall rocks. The blocks studied in this project were located in the 1-C orebody at T-3 Mine. Blocks in the 1-C are mined out transversely from hangingwall to footwall. The primary blocks have a strike length of 40 ft. (12 m) and the secondaries are 60 ft. (18 m) in length. The two study blocks were Block 19 and Block 21 between the 2850 and 2950 sublevels (see Figure 1.) and were both primary blocks.

The ore zone in Block 19 was 130 ft. (40 m) wide and in Block 21 it was up to 100 ft. (30 m) wide. The hangingwall rocks range from biotite schist to quartzite. The predominant footwall rock type is biotite schist. In both the hangingwall and footwall the biotite schist become increasingly siliceous away from the ore zone and eventually grade into quartzite 10 to

20 ft. (3 to 6 m) in the hangingwall and footwall. The hangingwall drift on 2850 sublevel is in silicious schist and quartzite with occasional amphibolite and pegmatite bands.

The rockmass characteristics were assessed by conducting line surveys in blocks 19, 20 and 21. The information was then processed to get an IRMR or Inco Modified Rock Mass Rating. IRMR is a modification of the RMR (Rock Mass Rating) method proposed by Bieniawski (1979). The IRMR was used to calculate the Rock Mass Elasticity and the material constants for the ore zone and wall rocks. Uniaxial compressive strengths (UCS) and Poisson's ratio are average values derived from lab tests that have previously been done at the Thompson Mine. Table 1 lists the rockmass properties. The m and s values are material constants that define a quadratic curve following Hoek's failure criteria (1980). Undisturbed values are more applicable to the open pit environment while disturbed values better reflect underground conditions. Joint alteration (Ja) and joint roughness (Jr) were measured directly during line surveys.

Table 1. Rockmass Properties

Parameters	Ore Zone	Hangingwall and Footwall schist
Uniaxial compressive strength, MPa	70	150
Inco Rock Mass Rating	72.29	73.96
Rock Mass Elasticity, GPa	36.08	39.72
Poisson's Ratio	0.25	0.17
Material constants: m (undisturbed) s	9.29 0.0460	3.95 0.0554
Material constants: m (Disturbed) s	3.45 0.0099	1.56 0.0130
Joint alteration, Ja	0.75 - 1.0	0.75
Joint Roughness, Jr	2 - 3	1.0

2.2. Experimental Drift

An experimental drift (Figure 2) was driven on the 2850 sublevel in the hangingwall rock in order to provide access for instrumentation installation purposes for blocks 19 and 21. The drift layout size is 10 ft wide by 10 ft high (3 m x 3 m) and was increased at both the north and south extremities to 13 ft wide by 13 ft high (4 m x 4 m) to accommodate both the diamond drill

and the in-the-hole (ITH) percussion drill. The total length of the drift is 150 ft (45.7 m). Block 19 and 21 drill rooms are located on each side of the experimental drift in the orebody. In each drill room, the 8-inch (20 cm) holes were used to insert the scan head of the Cavity Monitoring System. The plan view of the experimental drift and blocks 19 and 21, as well as extensometer holes, are also shown in Figure 2. Figure 3 shows the cross-section view of Block 19 and the relative position of extensometers.

The sole purpose of the percussion holes was to install the extensometers which monitor the ground movement of the hangingwall. The diamond drill holes (just beside the extensometer holes) on the other hand served many purposes. Firstly, the core from the holes was logged and samples were sent out for mechanical testing. Secondly, these holes provide a means by which the borehole camera can effectively view each hole and ascertain the degree of fracturing. Finally, the borehole dilatometer was used in these holes to determine the short term deformability of the rock.

Figure 4 shows a plan view of 2850 sublevel for instrumentation layout of cable strain measuring device (CSMD), vibrating wire and extensometers.

Figure 5 displays the location of the blast monitoring points. These points are numbered from 1 to 7. Points 1 and 6 are located in the hangingwall of block 20. Points 2 and 5 are located in the orebody on each side of 20 block access drift. Monitoring points 3, 4, and 7 are located in the footwall portion. If a blast was to occur in Block 19, then the accelerometers and geophones would be installed on points 1, 2, 3 and 7. Alternatively, if a blast was to occur in Block 21, the monitoring points 4, 5, 6 and 7 would be used. Other points shown in Fig. 5 were used to measure P-wave velocities.

2.3. Blasting

2.3.1. Block 19

Figure 6 shows the plan view of Block 19, indicating the positions of blastholes. The 2850 sublevel access drift and drill room is indicated by the solid line and the 2950 sublevel access drift and undercut is shown by the dashed line. Block 19 was mined transversely in two separate blocks named 19A and 19B. The hangingwall portion, Block 19A, was mined first and the footwall side (Block 19B) was slashed into the void created by Block 19A. Total ore tonnage for Block 19 was 61,500 short tons (55,800 metric tons) grading at 4.1% Ni.

Block 19A consists of 11 rows on a 4 feet (1.2 m) spacing. A total of 5461 feet (1664 m) of percussion drilling was required. Drill holes are 3.5" (89 mm) in diameter. There were a total of 34,650 short tons (31,430 metric tons) of ore in this block. A slot is created by slashing into a 42-inch (1.1 m) diameter bore slot/fill raise which was first used for backfill for the blocks below. The slot is gradually opened up the full 40 feet (12 m) length of the block by approximately 20 feet (6 m) in width. A 20 feet (6 m) crown pillar is left so that men may

load and blast and clean holes with the percussion drill. Blasting then proceeded towards the hangingwall. The final blast would typically include the last two or three rows closest to the hangingwall along with the crown pillar. When blasting rows parallel to the hangingwall, the toe burdens and spacings effectively become a 10 feet by 8 feet (3.0 m x 2.4 m) pattern. The explosives product used for this block was exclusively ICI's R9205V repumpable bulk emulsion. Block 19 A was mined out in 7 blasts.

Block 19B is the footwall portion of Block 19 and contained some 36,850 short tons (33,430 metric tons) of ore. There are 13 drill rows (Rows A-M) on an 8 feet (2.4 m) spacing. Total footage drilled was 6379 feet (1944 m) of 3.5" (89 mm) diameter holes. Block 19B was blasted in three separate blasts. All blasts were slashed into the void created by Block 19A. The initial blast consisted of rows A to D, while the second was rows E to H and the final was rows I to M. The first two blasts used the repumpable emulsion while the third blast used Aquamex 75 mm diameter cartridges.

2.3.2. Block 21

Figure 7 displays the top and bottom sublevels of Block 21: solid lines for the top sublevel and the dashed lines for the bottom sublevel with blasthole positions. There are 10 drill rows in this block. Rows A to E are on a 5 feet (1.5 m) spacing and rows F to J are drilled on a 4 feet (1.2 m) spacing. Toe spacings of holes was 16 feet (4.9 m). The blasting plan was to create a slot from rows F to J up to 20 feet (6.1 m) below bench elevation. Rows A to E were then slashed into the opening created by the slot. In total, 11 blasts were carried out to mine out this block.

Table 2 summarizes all the blasts in mining out the two blocks with the weight of explosives used and the measured VODs.

3. ANALYSIS OF RESULTS

This study mainly consisted of measuring damage to the hangingwall through instrumentation and differentiating between blast induced damage and stress induced damage. Direct measurement of damage assessment consisted of the borehole camera surveys in conjunction with the extensometer readings. Figures 8 and 9 for blocks 19 and 21 respectively show the fracture frequency observed from the borehole camera surveys and the displacement data obtained from the extensometers. The fracture frequency curves show the total number of fractures recorded from the surveys, the total number of open fractures and the percentage of open to closed fractures. The first borehole camera survey was performed on Feb. 08/94, a full month before any blasting had occurred in Block 19A. As shown in Figure 8, there were 20 fractures in total from the five observation holes with four fractures being opened. From blast 1 through to blast 5, the creation of 29 new fractures was observed for a total of 40 with only seven fractures being opened.

Table 2. Blast Summary

Blast number	Date	Number of hole	Weight of explosives, kg	Type of explosives	Measured VOD, m/s
19A - Blast 1	09-3-1994	2	290	R9205V	-
19A - Blast 2	11-3-1994	3	566	R9205V	5630
19A - Blast 3	14-3-1994	5	771	R9205V	5770
19A - Blast 4	15-3-1994	12	1,852	R9205V	5680
19A - Blast 5	18-3-1994	19	2,148	R9205V	5720
19A - Blast 6	21-3-1994	4	57	R9205V	5300
19A - Blast 7	28-3-1994	51	3,351	R9205V	5440
19B - Blast 1	06-4-1994	47	4,942	R9205V	-
19B - Blast 2	14-4-1994	49	3,876	R9205V	5320
19B - Blast 3	29-4-1994	46	2,034	Aquamex	4520
21 - Blast 1	12-4-1994	18	237	Amex II	3950
21 - Blast 2	19-4-1994	14	237	Amex II	3800
21 - Blast 3	04-5-1994	16	192	Aquamex	4500
21 - Blast 4	04-5-1994	14	167	Aquamex	-
21 - Blast 5	05-5-1994	14	157	Aquamex	4820
21 - Blast 6	06-5-1994	14	166	Aquamex	3440
21 - Blast 7	06-5-1994	12	166	Aquamex	4650
21 - Blast 8	09-5-1994	9 1	848 81	Amex II Tovan Lite	3350 4070
21 - Blast 9	10-5-1994	9 2	823 218	Amex II Aquamex	3830 -
21 - Blast 10	11-5-1994	6 8 2	694 75 108	Aquamex Amex II Tovan Lite	4830 - 4030
21 - Blast 11	13-5-1994	20 22 10	908 848 484	Aquamex Amex II Tovan Lite	- - -

The extensometer data confirms this fact showing very little movement occurring. Vibration levels recorded from blasts 2, 4 and 5 were in the order of 27-35 in/sec (686-889 mm/s) at distances of approximately 70 feet (21.3 m) and were considered quite high. No recordings were obtained from blast 3 due to equipment failure but it is thought that the vibration levels would have been in the same range as the others mentioned. Fractures were produced by these excessive vibration levels and are considered blast induced damage. Blast 6 was basically a non-event as only a few holes were blasted in the slot area with correspondingly low vibration levels in order to prepare it for the final crown/slash blast. Blast 7, the final blast, proved to be most damaging to the hangingwall. According to the CMS scan, the overbreak was as deep as 6 ft. (1.8 m) beyond the hangingwall contact. As well, 40% of the hangingwall holes were ineffective. The number of fractures increased 63% from 49 to 80 but, more importantly, the number of open fractures had a more than threefold increase, from 7 to 23. The extensometer data showed that the displacement was unusually large. The maximum displacement recorded was 1.4 inches (3.6 cm). Not only were the displacements large but many of the observation holes had offset. A total of 39 feet (11.9 m) of the observation holes were lost, thus preventing the use of the borehole camera over the full length of the hole. Vibration levels were in the order of 30 in/sec (762 mm/s) at a distance of 61 feet (18.6 m) in the hangingwall, confirming that this level of vibration is over the threshold which produces significant damage. The extensive damage to the hangingwall was believed to be caused by the close contact of blastholes against the hangingwall and the use of the powerful bulk emulsions.

There were three more blasts in Block 19A and as each blast got further from the hangingwall they produced lower and lower vibration levels. One week after the final blast in Block 19A, a borehole camera survey reported another 30 feet (9.1 m) of holes were inaccessible due to hole offsetting. The period covering from the final blast in Block 19A to the final blast in Block 19B shows that the number of fractures had increased from 80 to 141 with the number of open fractures increased from 23 to 52. The damage observed during this period was mainly due to stress redistribution as the void became increasingly larger. Although the displacements recorded were continually increasing, the initial blast damage was created from blasts 2 through 5 and 7 in Block 19A.

The numerical modelling results clearly show the large zone of low confining stresses along the unsupported span of the hangingwall once Block 19A is mined out. Zones of tension are equally present. A similar profile of the confining stresses is seen when block 19B is fully mined out.

The creation of fractures eventually ceased in late April/94 as indicated from the borehole camera surveys but the hangingwall was still moving as broken material was being mucked out from the block. Extensometer readings did not stabilize until the block was 75% filled with cemented rockfill in early July 1994.

Figure 9 displays the fracture frequency and displacement curves for Block 21. Very few fracture were reported in the initial survey of Block 21 on Feb. 08/94. There were eight fractures in total of which two were open. On April 08/94, before any blasting had occurred

in this block, a total of 51 fractures were reported of which 10 were deemed open. After the first two small crater blasts were taken, the total number of fractures had increased to 95 with the number of open fractures accounting for 20 of them. The vibration levels for each blast were in the order of 8 in/sec (203 mm/s) at 65 feet (19.8 m) in the hangingwall. It is thought that the creation of these fractures were mainly due to stress redistribution, notably the influence of the increasing void of Block 19. It is unlikely that these low vibration levels generated so many fractures. Also the amount of fracture opening was very little. The extensometer data showed minimal displacements for all extensometers.

The total amount of fractures tended to stabilize thereafter increasing in only very small increments. There were five more crater blasts with little change in the amount of fractures. The highest vibration levels obtained was in blast 4 with a reading of 10.6 in/sec (269 mm/s) at a distance of 60 feet (18.3 m) at the hangingwall monitoring point. The last four blasts were somewhat larger slash blasts with correspondingly higher vibration levels. The highest vibration level recorded from these last four blasts at the hangingwall monitoring point was 14.8 in/sec (376 mm/s) at a distance of 64 feet (19.5 m). The general tendency of the total number of fractures remained flat, however the total number of open fractures kept increasing at a moderate rate. The amount of displacements remained quite small and only increased significantly after the final blast was taken when the unsupported span was at its largest. All of this data suggests that the damage to the hangingwall in Block 21 was more stress induced in nature as opposed to being blast induced. Therefore the level of vibration produced from these blasts would fall into the acceptable range where little or no blast damage can be induced. Only five feet (1.5 m) of the observation holes had offset to prevent the borehole camera from gaining access. A total of 16 feet (4.9 m) of slough was reported from the observation holes after the final blast but this slough resulted mainly from undercutting the hangingwall and was not blast related. The displacements kept increasing after the final blast as muck was being drawn out of the block. Filling the block started on June 07/94 and by July 10/94 it was 60% complete. Extensometer readings closest to the hangingwall stabilized at this point.

The direct measurements of damage indicated that some blast induced damage to the hangingwall was evident during the blasting of Block 19A. Further analysis into the final blast reveals the following.

The final blast in Block 19A used 7388 lbs (2251 kg) of the repumpable bulk emulsion with a rated VOD of 5500 m/s. This blast consisted of the last two to three rows parallel to the hangingwall and the crown portion. There were 15 holes in all that were considered hangingwall holes. Of these 15 holes, one third of them did not generate vibration traces and were deemed missed. The holes were AA1, B1, D1, F1 and I1 and it was thought that the ground fell before these holes detonated (cut-offs). As well, hangingwall hole J1 produced very low vibration levels and did not contribute to the breaking of ground. This amounted to 40% of the hangingwall holes being ineffective. If one considers how the hangingwall was shaken up after this blast as recorded by the borehole camera surveys and the extensometer data while keeping in mind that there were six misfires in the hangingwall holes, the explosive used must be too powerful or in other words contain too much energy. The high ratio of misfired

hangingwall holes could be another indication of blast damage in the vicinity of blastholes with bulk emulsions, exhibited as overbreak and cut-offs to nearby blastholes. The hangingwall stope boundary requires constant special attention in order to minimize blast damage by utilizing proper lower density explosives in hangingwall holes and production holes adjacent to the hangingwall holes, therefore reducing unwanted hangingwall dilution.

The hangingwall stope boundary was first mentioned because of the emphasis placed on it over the course of this study due to the important economic issue that dilution raises. The analysis should not be limited to that boundary alone. Other stope boundaries are just as important. The north and south boundaries of a block or pillar walls are other examples. Sloughing occurred on the south ore wall of Block 19 between May 5/94 and June 10/94. This period of time covers from the final blast in Block 19B to when filling of the block was about to commence. The approximate limits of this ore sloughing was from the 11040 East section to the 11100 East section and from the 7820 elevation up to the 7900 elevation. The blasts that cover this area are blasts 4, 5 and the final blast in Block 19A and the first in Block 19B. The sloughing was considered so serious that mucking of the block was suspended and a large amount of broken material estimated at 10,000 tons (9070 mt) was left behind. The damage to the south wall was seemingly blast induced although there is only indirect measurements of vibration traces available. The north and south stope boundaries were not monitored to the same extent as the hangingwall was but there were high vibration levels originating from many holes in that area. Not helping the situation is the stress redistribution around the newly created cavity. Figure 10 shows a plan view of the minor principal stresses around Block 19. Zones of tension are created on both pillar walls after Block 19B is mined out indicating areas that are more susceptible to damage.

It is important to keep vibration levels as low as possible. The three most important sources contributing to high vibration levels are the weight of explosive per delay, the degree of confinement of the explosive charge and the timing between the explosive charge. Table 3 lists the results from some selected holes in Block 19A which were all single events.

While reducing the charge weight per delay is a good general guideline to follow in helping reduce vibration levels, it is not the only factor to be considered. The confinement of the charge should also be taken into account. This factor is presented in Table 3 by the toe break burden and the toe break angle. The general tendency is the larger the burden, the higher the resulting vibration level. Also, higher vibration levels will be obtained as the break angle narrows. That is, the less free face an explosive charge has, the more vibrations will be transmitted to the surrounding rockmass.

Referring to Table 3, holes C6, J4 and K5 of blast 4 all produced relatively high vibration levels. All holes had high charge weights per delay and fairly narrow break angles. The lowest vibration level was obtained from hole K5 which had the smallest toe burden to break. The highest vibration level was obtained from hole J4 which had the narrowest toe break angle.

Table 3 Summary of High Vibration Level in Correlation with Toe Break Burdens

HOLE	BLAST	EXPL.	WT. (kg)	DIST. (m)	PPV of vector sum (mm/s)	BURDEN (m)	ANGLE (°)
C6	4	EMUL.	166	25.9	498	2.4	65
J4	4	EMUL.	170	22.3	699	2.4	32
K5	4	EMUL.	189	23.8	526	2.0	68
F4	5	EMUL.	149	19.2	851	2.3	54
G3	5	EMUL.	137	20.4	831	3.0	62
K3	5	EMUL.	161	25.0	249	2.0	123
F5	6	EMUL.	17	17.4	302	0.5	57*
E1	7	EMUL.	79	13.4	345	2.4	48
E3	7	EMUL.	144	15.5	353	2.4	112

* in raise.

Among the holes F4, G3 and K3 of blast 5, hole K3 generated the lowest vibration level and was the hole with the lowest burden and highest break angle, yet it contained a slightly higher explosive charge.

Hole F5 in blast 6 demonstrates how difficult it is to break into a circular raise. Only 37 lbs (17 kg) of explosives were used but this charge still generated vibration levels of 11.9 in/sec (302 mm/s) at a distance of 57 feet (17.4 m). The burden was only 1.5 feet (0.5 m) and the break angle was only 57 degrees.

Holes E1 and E3 of blast 7 show that similar high vibration levels were obtained even by reducing the charge weight by half because the break angle was significantly lower. Hole K2 and K5 show that higher vibration levels were obtained with a lower charge weight, larger burden and narrower break angle.

Investigating the effects of the charge weight per delay, the toe break burden and the toe break angle is critical for every blast. The optimization of these factors will lead to more efficient blasting practices.

4. CONCLUSIONS AND RECOMMENDATIONS

The comprehensive studies on blast damage in this project has generated a voluminous database which contains recording results from many types of instrumentation. The correlation between the results of borehole camera, extensometers, vibration monitoring and numerical modelling enabled differentiating between blast induced damage and stress induced damage. The CMS measured voids accurately and enabled calculation of dilution from overbreak and sloughing.

It is concluded that evident blast damage was produced to the stope boundaries including the hangingwall during the mining of Block 19A. This damage produced operational problems such as mucking delays and secondary breaking. The rockmass damage measured on the hangingwall side of Block 21 was more stress induced in nature. It is recommended to take steps to reduce both blast induced and stress induced rock damages.

From the observations, one can see that blasts can produce damage not only to the immediate rockmass but also increase rock damage by extending the existing fractures. Considering overall rockmass damage which contributes to ore dilution and rock instability, one can divide the damage into three categories:

(1). Natural fractures such as joints, faults and bedding planes, which exist before excavation.
(2). Fractures created by stress redistribution for zones subjected to tensile stresses;
(3). Damage by blasting can be further broken down into:
    (a) by shock energy which creates radial cracks around a blasthole and
    (b) by gas energy which extends and wedges into every fracture created by all the above factors.

The rockmass damage existing prior to the excavation is beyond the control of the mine operator. However, one can characterize the rockmass for adopting different strategies to excavate it. Damage due to stress redistribution can be coped with by proper design of block size and mining sequence. For blast damage control, one needs to reduce the effects of both shock energy and gas energy. In practice, the following methods can be used to reduce blast induced damages:

(1) to use decoupled charges;
(2) to use low shock energy explosives and
(3) to design a blast pattern and initiation sequence to reduce the effective burden and to enlarge break angles.

The occurrence of blast induced damage to open stope boundaries clearly indicates the need to apply controlled blasting techniques to help minimize this type of damage. It is recommended that mining engineering emphasize open stope perimeter control in order to reduce dilution and minimize operational problems by utilizing the CMS to measure the voids created,

measuring vibration levels to obtain site specific vibration attenuation curves and adjusting the blast design. Blast induced damage can be controlled by selecting lower density explosives on the two rows closest to a stope boundary. The bulk emulsion used in this study was too powerful for perimeter holes. Blasting crews must be trained and encouraged to use low energy explosives for perimeter holes.

It is recommended to train blast designers and the blasting crews the principles of blast design, specifically the perimeter control aspect, the influence of burden and break angles on vibration levels and their effects on damage and proximity effects. Blasting crews must respect the blast designs. Although this is a question of management and training, production can be significantly affected by malpractice such as improper designs or, if the design is correct, it is not properly followed.

The major factors that result in stress induced damage are the size and duration of opening of an excavated block. Since the size (length, width and height) of a block must follow the general mining plan, it is recommended to shorten the excavation period as much as feasibly possible. If the ground stress becomes extremely high, the dimensions of the blocks must be modified to reduce dilution and fracture development in the surrounding rockmass. For the blocks which have been excavated over a certain period of time, the final few blasts should be carried out with special care because at this point, fractures have developed in some stressed areas. The gas penetration into these fractures could simply enhance stress induced damage. Therefore, for the crown pillar and hangingwall holes, controlled blasting should be a mandatory practice.

Numerical modelling based on CMS scan results should become common practice in mines in order to make a preliminary assessment of potential damage to the rockmass. The blast design for highly stressed blocks should be directly aimed to reduce blast damage following the principles discussed above.

ACKNOWLEDGMENT

This project was co-funded under the Canada/Manitoba Mineral Development Agreement with INCO Manitoba Division. The Mines Research Department of INCO Manitoba Division carried out the research and cooperated closely with the Scientific Authority from CANMET. The authors wish to thank the following major participants in INCO Mines Research: John Hilchey, Glenn Ludwig, Les Mansell, Tony Mansell, Al Meston, Dave Nicholls and Rob Smith. The Mine Operations and Engineering personnel of T-3 Mine are appreciated for their support during the course of this project.

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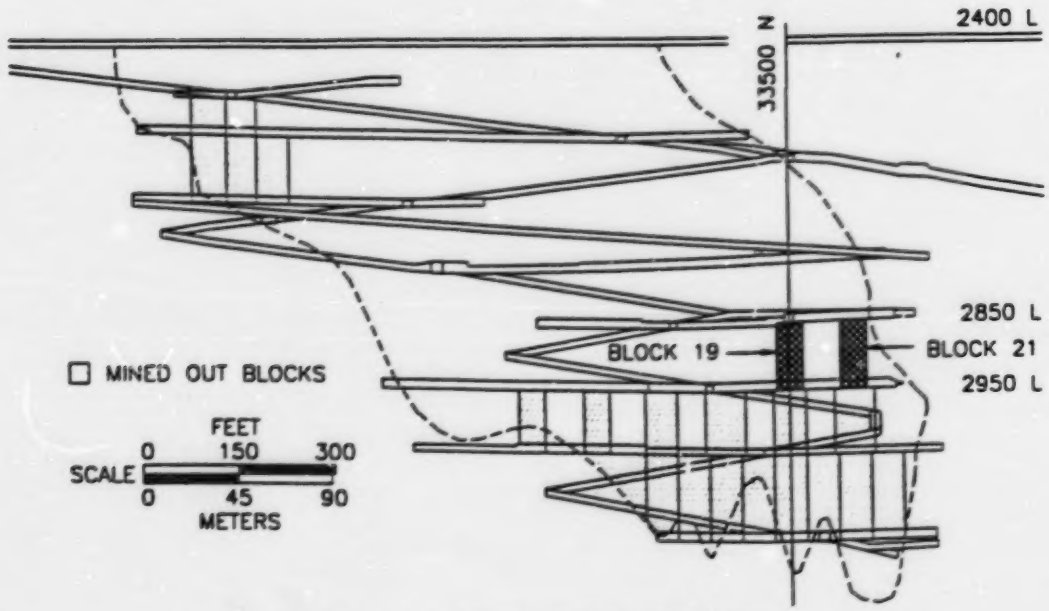


Figure 1. 1-C Longitudinal of T-3 Mine

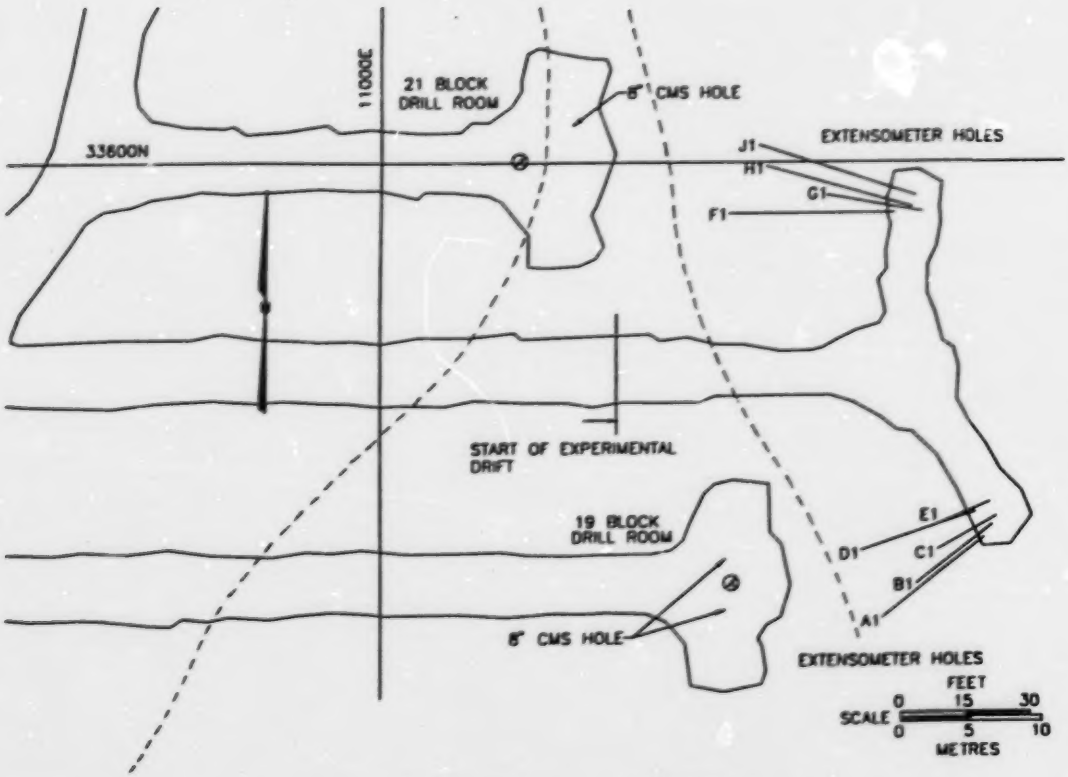


Figure 2. Experimental Drift Layout

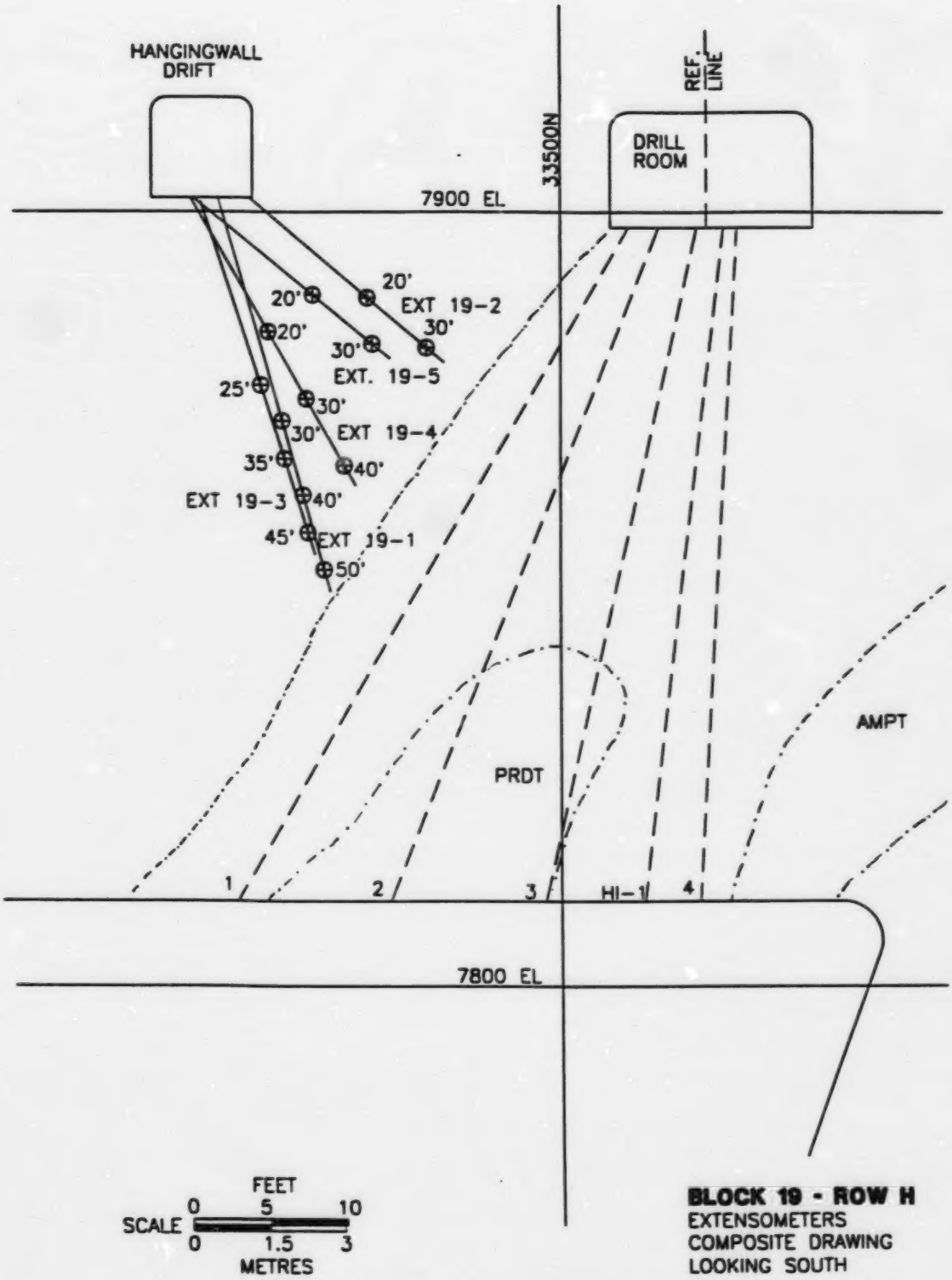


Figure 3. Block 19 Extensometer Locations

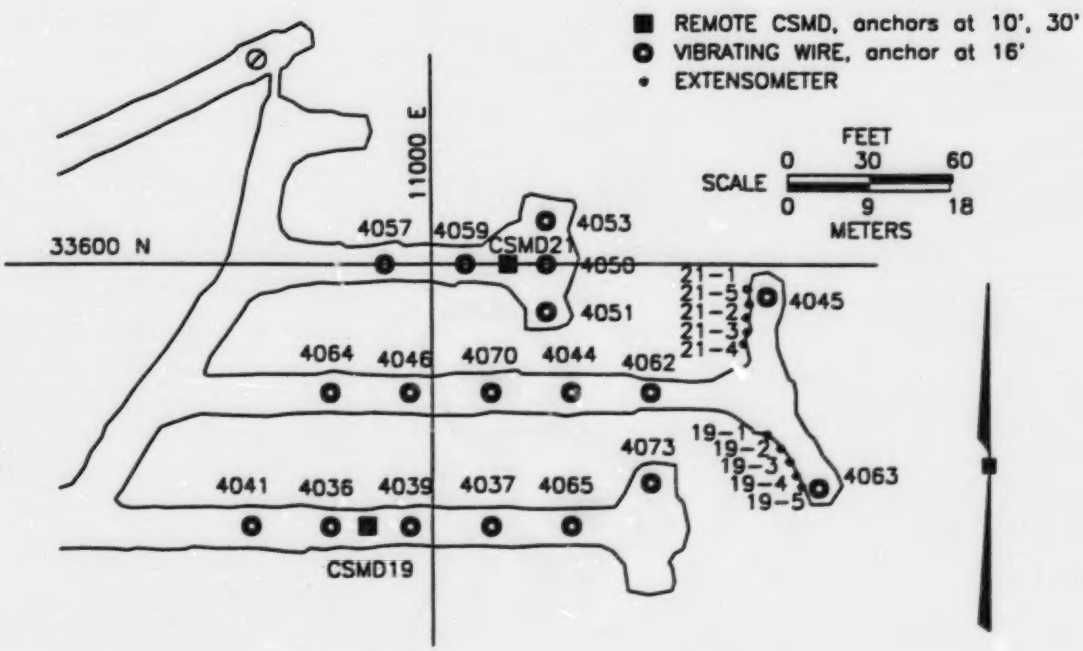


Figure 4. Geomechanical Instrumentation Layout

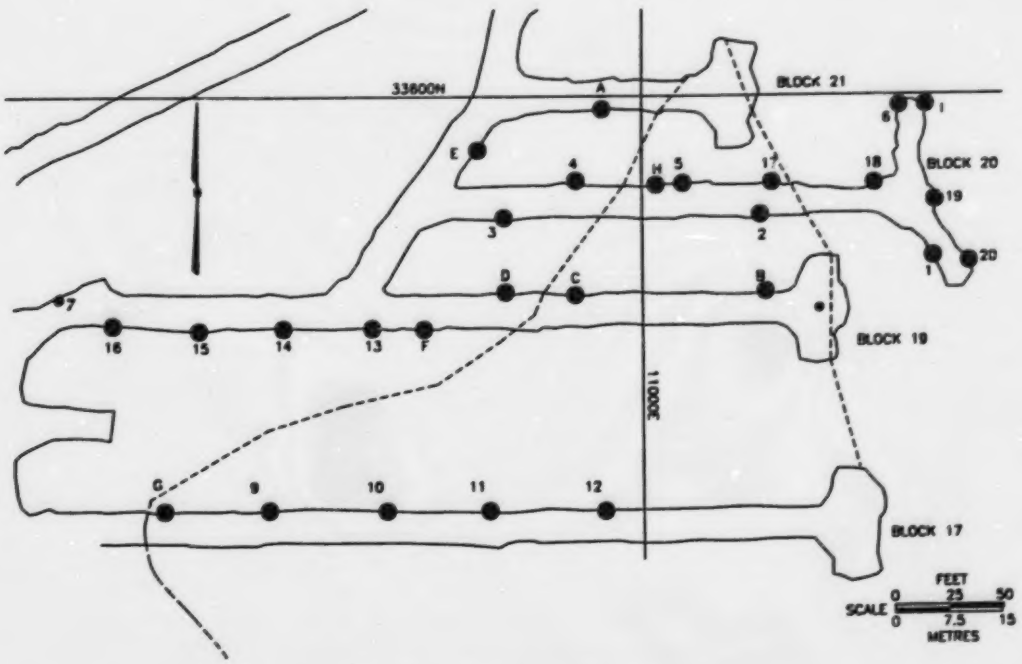


Figure 5. Blast Vibration Monitoring Point Locations

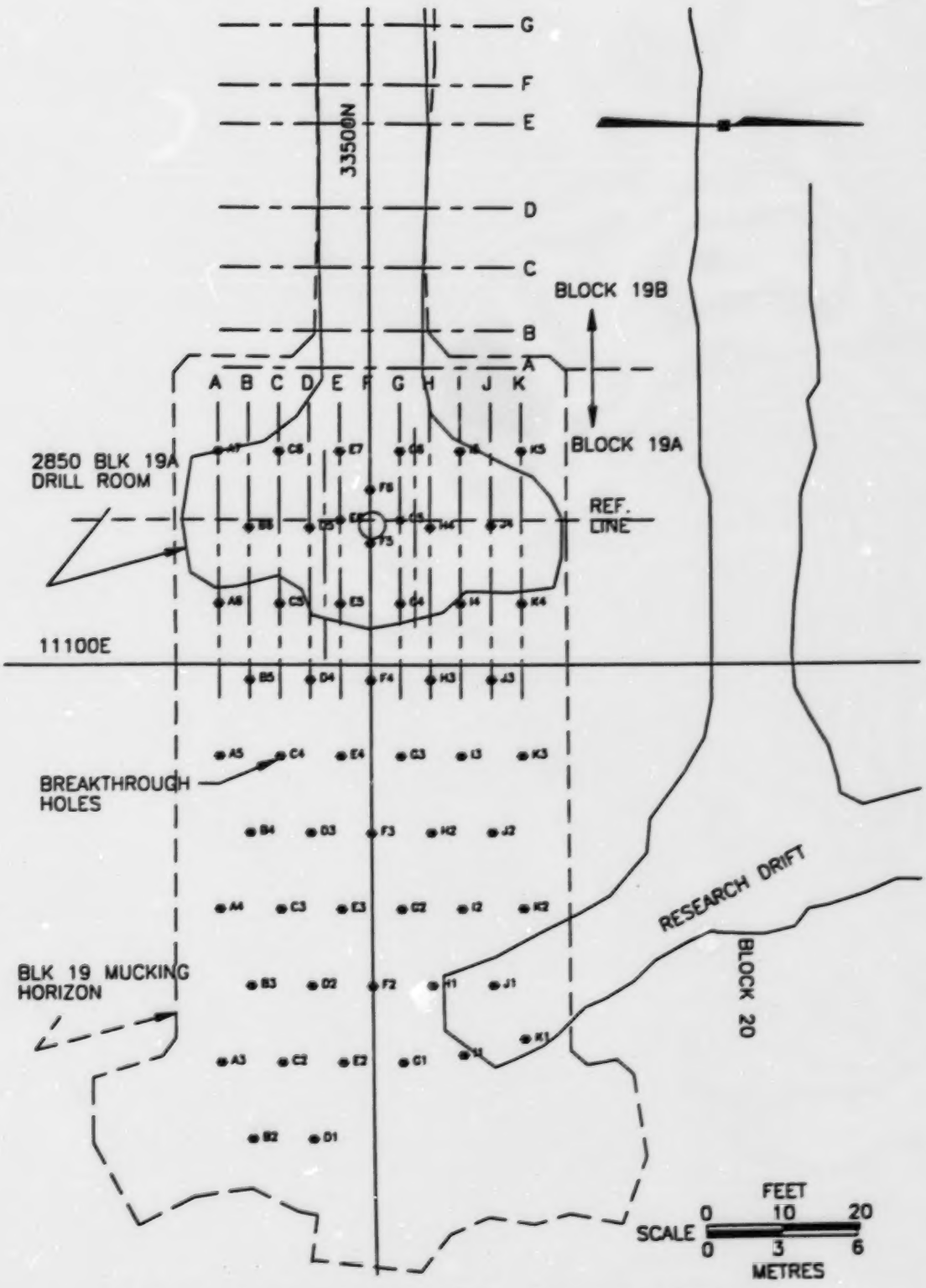


Figure 6. Composite Plan for Block 19: Location of Blastholes

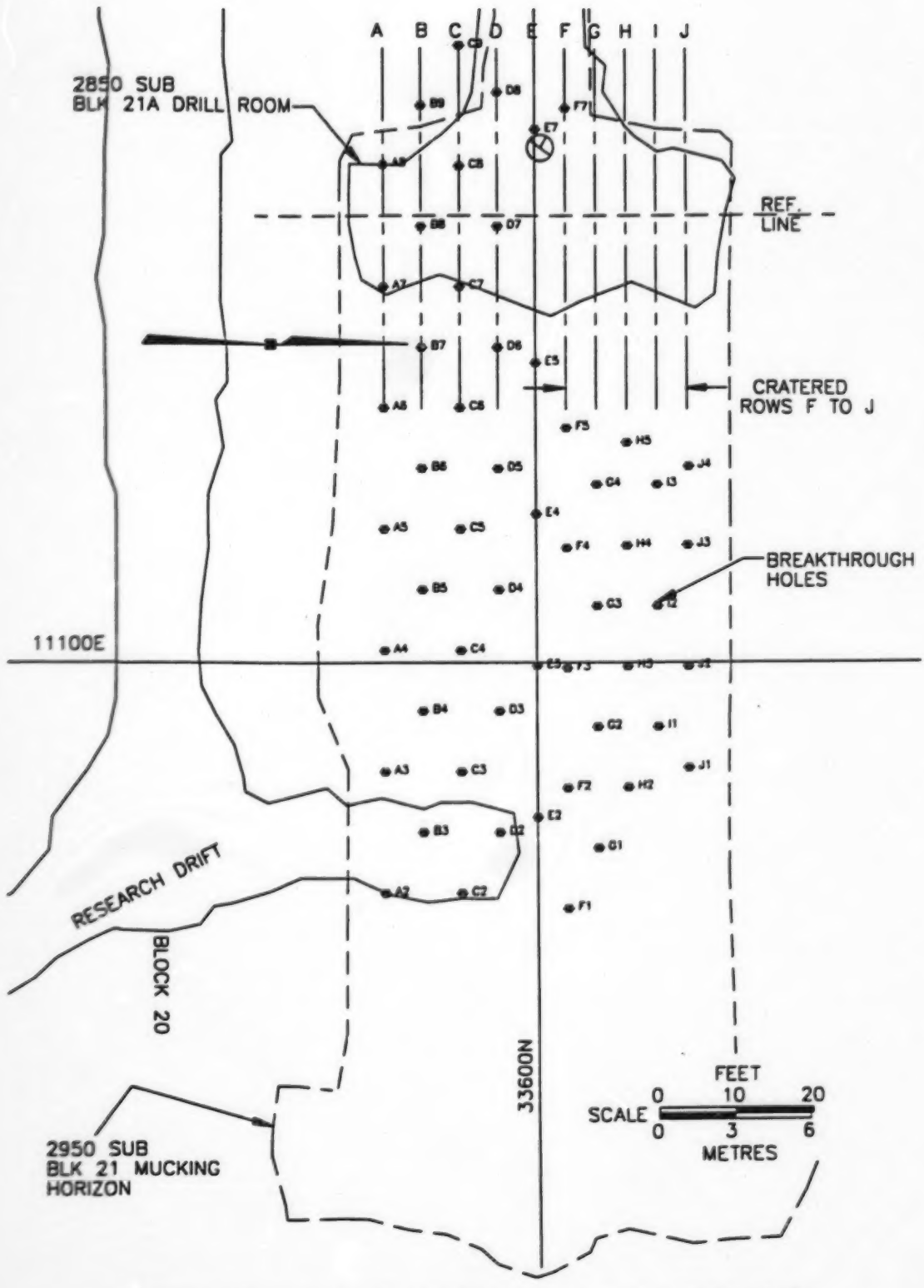
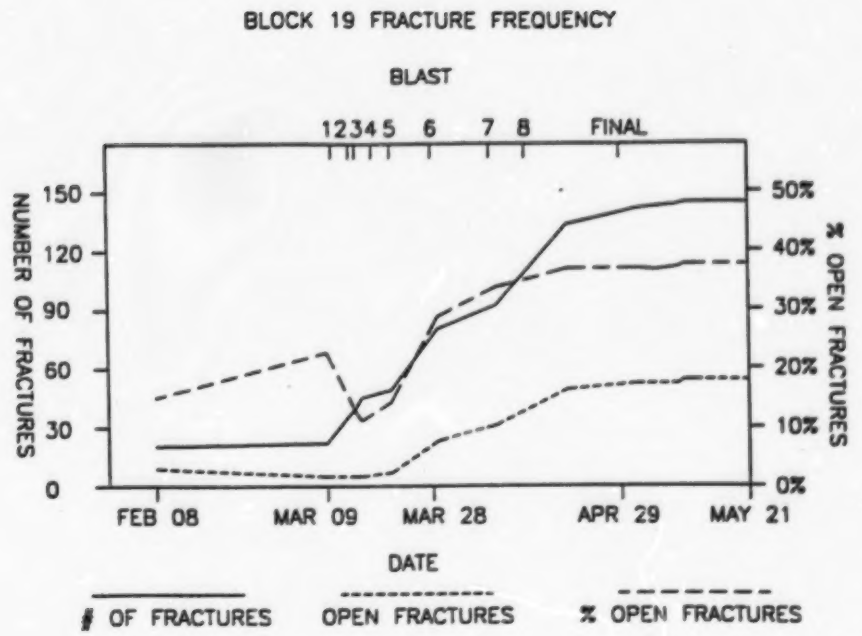


Figure 7. Composite Plan for Block 21: Location of Blastholes



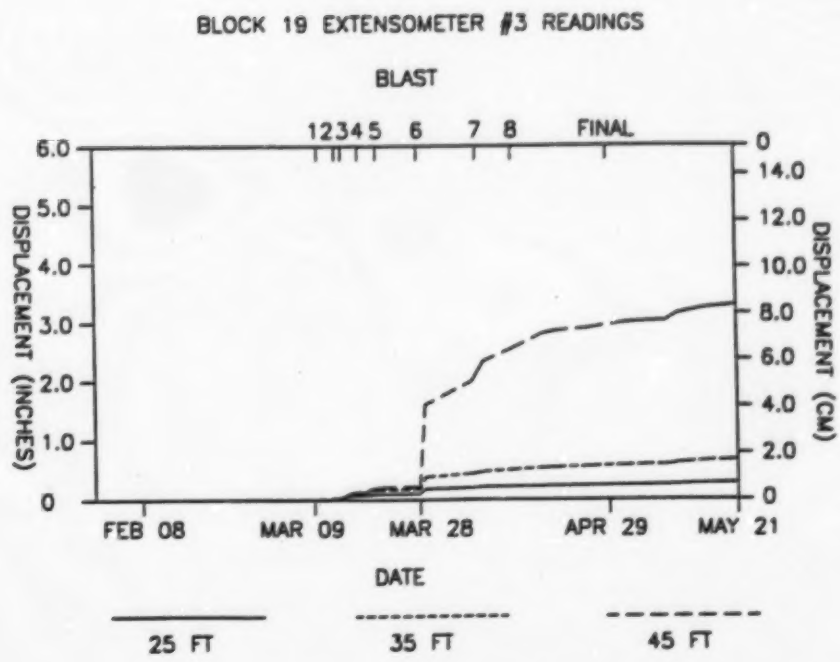
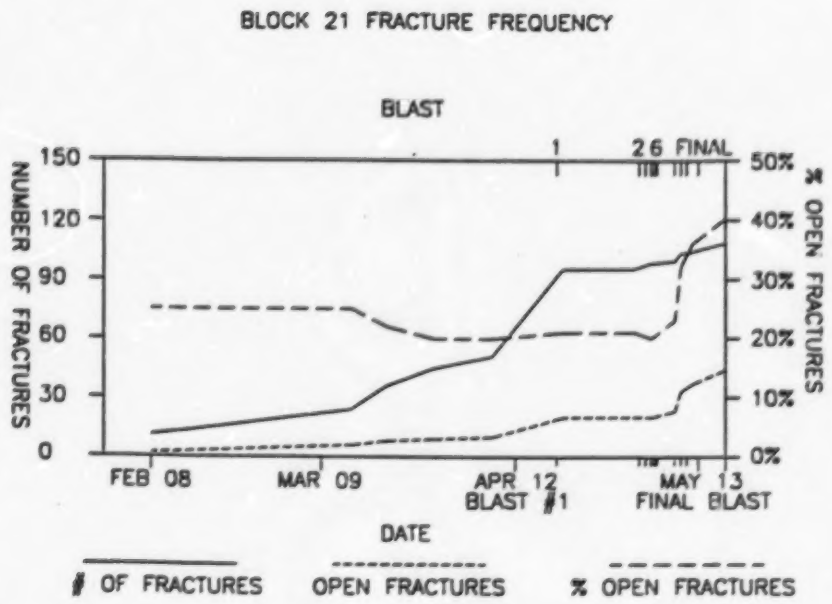


Figure 8. Block 19 Borehole Camera and Extensometer Results



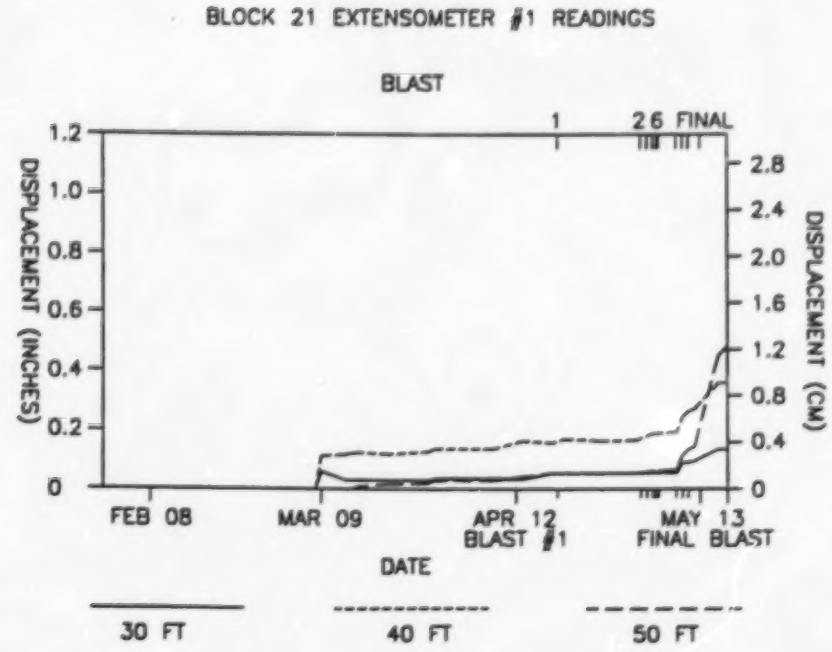


Figure 9. Block 21 Borehole Camera and Extensometer Results

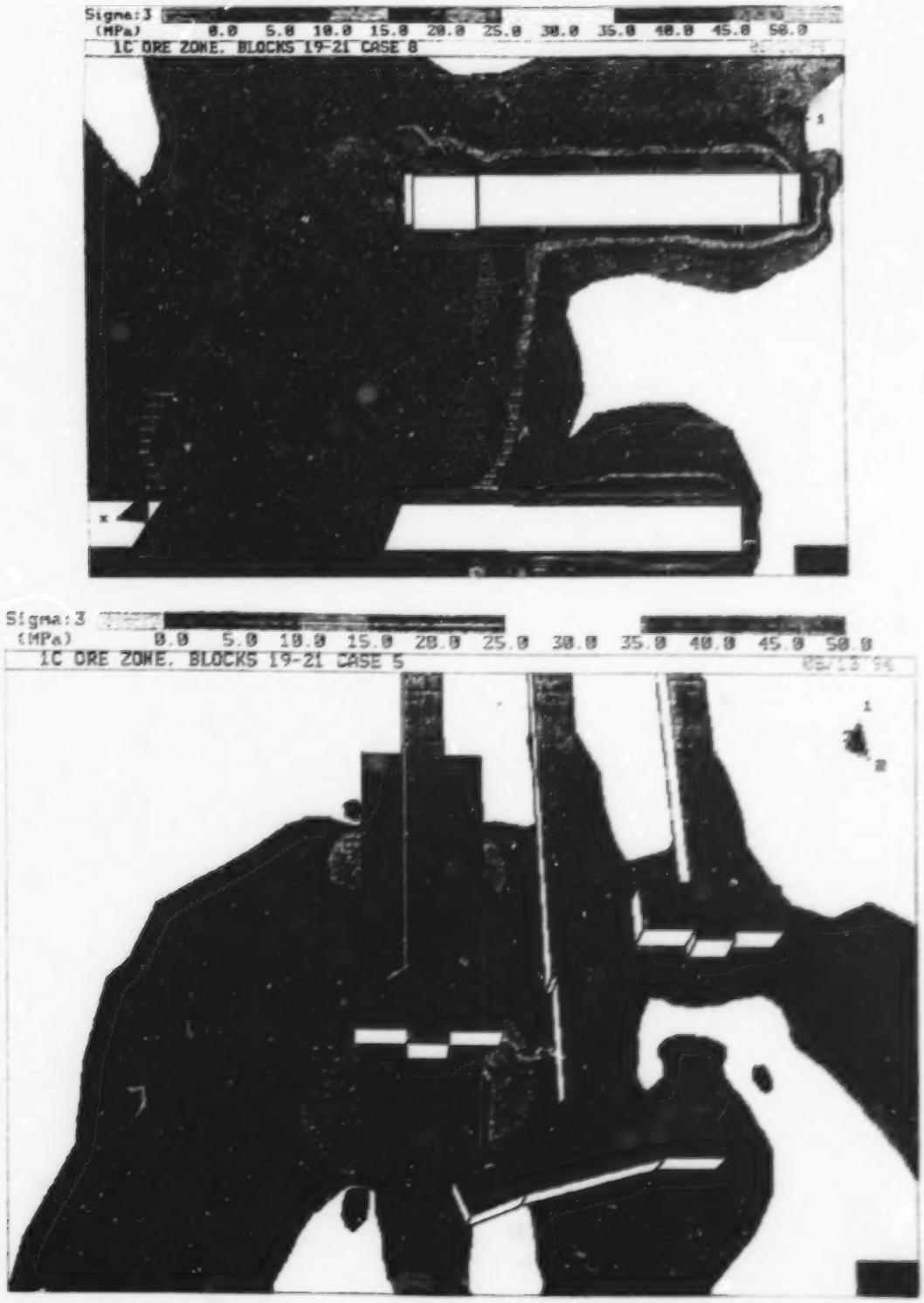


Figure 10. Block 19 Minor Principal Stresses